BLOCK CAVING IN LIMESTONE AT THE CRESTMORE MINE, RIVERSIDE CEMENT CO., RIVERSIDE, CALIF.

BY ALBERT E. LONG AND LEONARD OBERT

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BLOCK CAVING IN LIMESTONE AT THE CRESTMORE MINE, RIVERSIDE CEMENT CO., RIVERSIDE, CALIF.[1]

by

Albert E. Long[2] and Leonard Obert[3]

SUMMARY

This paper presents a description of the operational mechanics involved in block-caving a granodiorite-capped, limestone deposit. This method was used for 24 years at the Crestmore mine, near Riverside, Calif., which was operated by the Riverside Cement Co. as a source of raw material used to manufacture a portland cement.

The Crestmore mine is the only place where a block-cave method has been used to mine a limestone deposit. Structurally, limestone is relatively strong, and it is not usually considered to be a substance amenable to mining by block caving. It is therefore a credit to the engineering department of the Crestmore mine that it developed a satisfactory, relatively efficient method of block-caving a structurally stable rock.

In the discussion, covering the methods applied to produce about 7.9 million tons of limestone and granitic rock materials by block caving, particular emphasis has been placed on an assessment of (1) physical properties of the deposit, (2) ground pressure and support, (3) methods used to induce fragmentation and maintain cave action, (4) surface and subsurface subsidence, (5) secondary blasting of oversize, (6) production rate, (7) safety, and (8) the advantages and disadvantages of the cave method as applied to this specific limestone deposit.

INTRODUCTION

Between 1930 and 1954, 7,882,000 tons of limestone and granitic materials were extracted from the Crestmore mine by block-caving methods. Three reports[4 5 6] describing the deposit and various aspects of the mining operation have been published.

1/ Work on manuscript completed April 1957.

2/ Mining research engineer, Division of Mineral Technology, Bureau of Mines, Region II, Reno, Nev.

3/ Chief, Applied Physics Laboratory, Bureau of Mines, College Park, Md.

4/ Robotham, C. A., Mining Limestone by a Caving Method at Crestmore Mine of the Riverside Cement Co., Crestmore, Calif.: Bureau of Mines Inf. Circ. 6795, 1934, 20 pp.

5/ Wightman, R. H., New Caving Procedures at the Crestmore Limestone Mine: Trans. AIME, vol. 163, 1945, pp. 215-224; Mining Technology, vol. 8, No. 6, 1944, 10 pp.

6/ Bucky, Philip B., Block Caving at the Crestmore Mine: Explosives Eng., vol. 23, No. 3, May-June 1945, pp. 108-110, pp. 114-124.

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In 1954 and 1955 an open-stope, room-and-pillar mine was developed and placed in production in 1956. The unsupported rooms are 60 feet wide and 200 feet or more long. The current room-and-pillar mine and methods have been described by Wightman.7/

The Crestmore mine is the only known example of block caving used to mine a limestone. The fact that the same deposit, at a lower horizon, is now being open-stoped is an indication of the strength of the limestone. Structurally, it is one of the most competent rocks to be block-caved.

In preparing this report, most of the data have been obtained by reconstruction using detailed mine plans, cross sections, and production records as a basis. Details regarding the initiation and maintenance of caving in the blocks and the associated surface subsidence were obtained from interviews with the engineering staff of the mine.

ACKNOWLEDGMENTS

The authors are grateful to R. H. Wightman, superintendent of mining; Frank Brown, supervisor of mining; Peter Nalle, assistant mine superintendent; and C. D. Chandler, chief mining engineer, Riverside Cement Co., for their generous assistance in collecting the data for this report.

GEOLOGY

Limestone deposits in the Crestmore area are probably remnants of Carboniferous8/ formations that have been intruded by or enfolded in Jurassic granitic rocks. Since Jurassic time the limestones and associated intrusives have been tilted, faulted, and eroded. Most of the limestone deposits are blunt-edged, thick lenses. Some extend to a considerable depth down dip.

The Crestmore mine deposit consists of two lenses of metamorphosed and recrystallized limestone separated and surrounded by an igneous intrusive (granodiorite). North-south and east-west sections of the lower (Stanley) bed or lens are shown in figures 1 and 2. The beds strike north-south and dip approximately 50° E. The strike length of the lower bed is approximately 2,500 feet. It is 270 feet thick near the center, tapering to a 100-foot thickness near the lateral limits.

Metamorphism has almost obliterated the original bedding in the limestone. A remnant system of bedding is indicated principally by variations in mineral composition and by abrupt changes in the crystal pattern. The few true bedding planes that remain persist for only short distances and are generally tight and healed. These few planes are so tightly healed that drill cores seldom break along them, and long pieces can generally be recovered even when the core has been cut perpendicular to the bedding planes.

Sporadic zones of jointing exist in the limestone and, like the bedding, they are generally tight and healed.

Solution channels in the limestone, ranging from minute cracks to large open cavities, are numerous near the surface. With depth, they diminish in size and number. A few have, however, been found at a down-dip depth of over 300 feet. The

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7/ Wightman, R. H., New Underground Mining at Crestmore: Min. Cong. Jour., vol. 42, No. 12, December 1956, pp. 33-36.

8/ Logan, Clarence A., Limestone in California: California Jour. Mines and Geol., vol. 43, No. 3, July 1947, p. 270.

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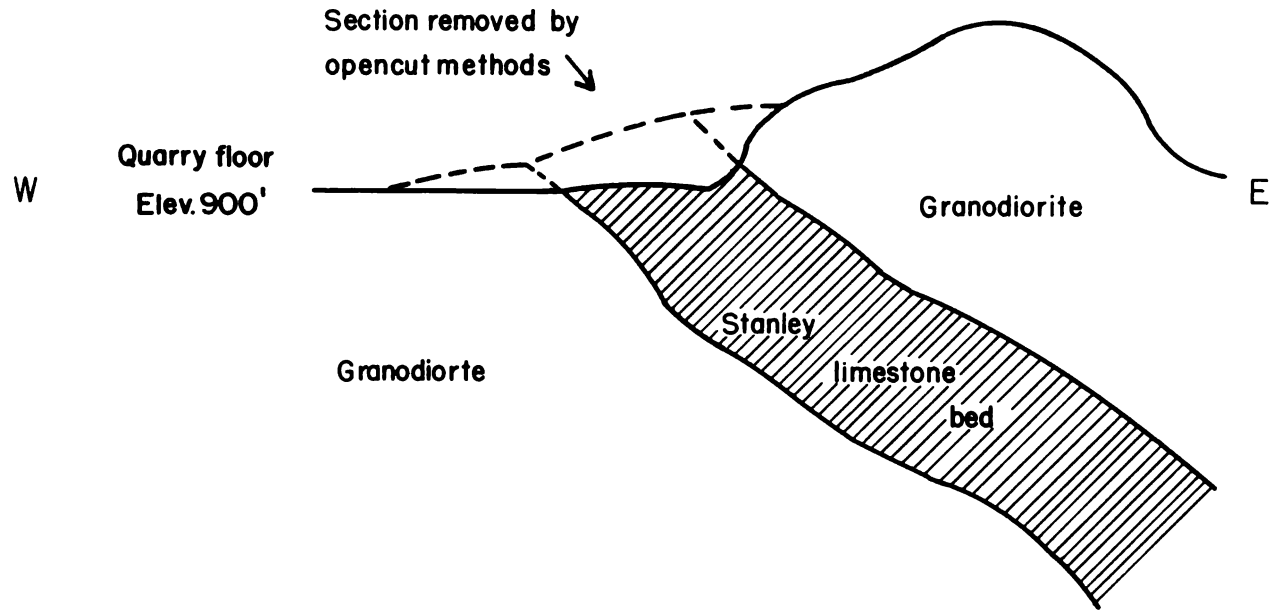

FIGURE 1. - East-West Section Through Deposit, Showing Area Removed by Opencut Methods.

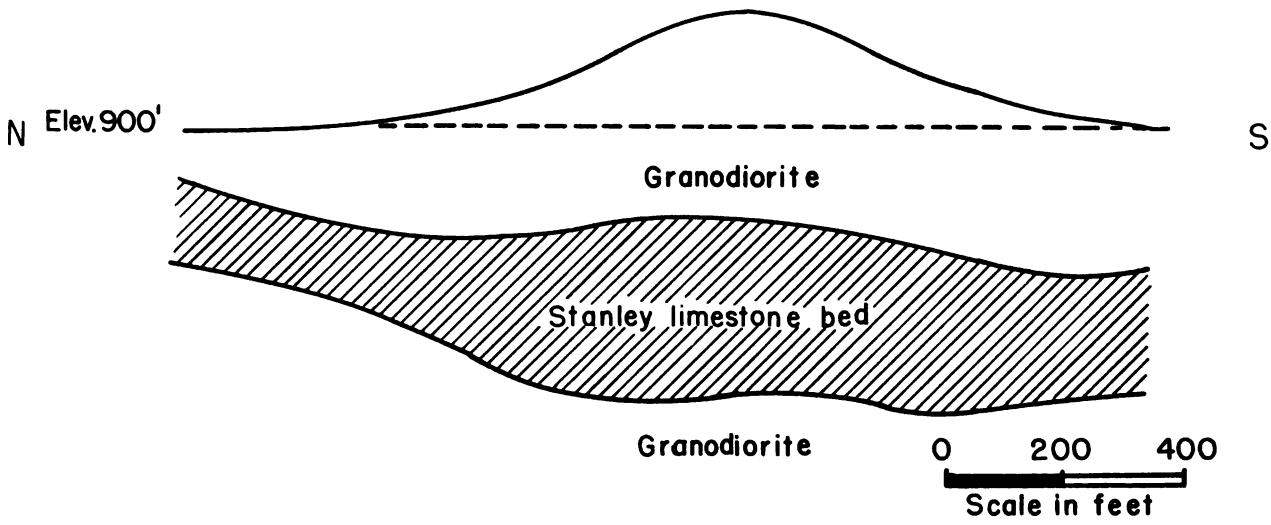

FIGURE 2. - North-South Section Through Deposit.

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solution channels tend to follow the general direction of preexisting joint planes. Many are partly or completely filled with decomposed rock materials and with solution products. There are enough solution channels in the upper 200 feet of the limestone to weaken the formation structurally. Hence, these channels are believed to have had a significant effect on the initiation and maintenance of caving.

The limestone is a white, fine- to coarse-grained, crystalline rock. The fine and coarsely crystalline facies of the deposit are usually stronger than the sugar-like, medium-grained varieties. Average magnesia-silica content of the limestone is low. The relatively high magnesia facies are usually finer grained and stronger than those containing less magnesia.

The intrusive rock forming the hanging and foot walls of the limestone lens is a gray, coarse-grained quartz diorite (granodiorite), with occasional hornfels inclusions. The quartz-diorite is cut by joints spaced from a few inches to several feet apart. Fractures are rare and widely spaced. At 200 feet below the surface the fractures appear to be tight. AX-size core, up to 5 feet in length, has been obtained in and below this horizon in both the foot- and hanging-wall rock. Near the surface the diorite is weathered, as indicated by staining and claylike fill material in the fractures.

An irregular zone of contact alteration not over 15 feet thick occurs in the quartz-diorite directly above the upper surface of the limestone. Scattered within the first 5 feet of this zone nearest the limestone-granodiorite contact are layered and irregular-shaped, iron-stained blocks of hornfels. The thickness, width, and length of the hornfels blocks vary from 1 to several feet. In some areas the hornfels masses are in contact with the limestone and in other areas they occur as inliers.

PHYSICAL PROPERTIES

The physical-property data in table 1 were obtained from diamond-drill cores taken from six blocks of limestone. Each block was selected as a representative sample of one of the various types of limestone in the deposit.

TABLE 1. - Physical-property data

Specimen No.	Specific gravity 1/	Compressive strength p.s.i.	Modulus of rupture, p.s.i. x 1,000	Young's modulus, 2/ p.s.i. x 10^6	Modulus of rigidity, p.s.i. x 10^6	Young's modulus, 3/ p.s.i. x 10^6	Scleroscope hardness, scleroscope units
1	2.68	7,700	2,170	9.10	4.02	9.02	42
2	2.72	7,900	-	6.58	3.15	6.8	39
3	2.71	8,500	2,260	11.1	4.17	11.0	45
4	2.68	11,000	2,300	9.82	4.11	5.82	44
5	2.70	6,700	-	2.41	2.29	5.82	39
6	2.72	24,000	2,360	11.4	4.37	11.3	56

1/ Apparent.
2/ Sonic.
3/ Static.

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Petrographically, the six specimens can be broadly classified as recrystallized carbonates. More specifically, specimens 1, 4, 5, and 6 were calcium carbonate (calcite) marbles; and specimens 2 and 3 were calcium-magnesium carbonate (dolomite) marbles. The principal physical distinction in these samples is in grain size. Specimens 1, 2, 3, and 4 were fine-grain marbles with 50 percent or more of the crystals less than 0.5 mm.; sample 5 was a medium-grain size with 50 percent or more of the crystals less than 1.0 mm.; and specimen 6 was a coarse-grain marble with crystals ranging from 1.25 to 15.0 mm.

The compressive strength of the samples ranged from 6,700 to 24,000 pounds per square inch. The modulus of rupture (outer fiber tensile strength) ranged from 2,170 to 2,360 pounds per square inch; hence, these marbles can be physically classed as relatively strong.

The physical properties of the granodiorites have not been determined. However, the granodiorites appear to be weaker than a true granite as a considerable portion of the hanging-wall granodiorite broke or crushed into a minus-1/4-inch material as the blocks were caved. The hanging-wall material, which did not crush easily, usually contained large amounts of hornfels.

GENERAL MINING METHODS

The principal mine levels and their vertical placement in relation to the lower Stanley bed and their elevation above sea level are shown in the generalized sections in figures 3 and 4. The system and pattern of drifts and tapping and transfer raises used throughout the operation at Crestmore were similar to those employed in block caving the porphyry-copper deposits at Ray[9/] and in the Globe-Miami district in Arizona. However, the physical properties of the material and the size of the blocks mined at Crestmore necessitated some modifications of the mining scheme as applied at Ray. These modified caving procedures and experience resulting from their application to caving limestone are of particular interest because they increase the range of ore strengths to which block caving may be applied.

The principal differences at Crestmore that necessitated a change from the Ray procedure were the type of rock and the depth of the cover. Although the Crestmore limestone contained a network of watercourses and accompanying areas deteriorated by solution, the rock did not contain, to any significant degree, the fracturing and jointing normally characteristic of block-caving ground. The cover overlying the limestone was thin, ranging from 0 in some areas to about 375 feet overlying block 4C. This maximum depth of cover affected only a small percentage of the total area caved. Hence, the weight on the blocks and correspondingly the stress in the limestone was slight, a factor that makes caving more difficult.

To induce and sustain caving in the ground it was necessary to effect measures that would concentrate the stress generated by the weight of a thin cover on localized areas near the lower surface of the blocks. These measures were either unique to this operation or were carried out to a degree greater than was required in normal caving. They were as follows:

1. The blocks were completely cut off on all sides by either vertical shrinkage (cutoff) stopes or by caved areas. The cutoff stopes extended from the mining and grizzly levels upward to the quarry floor or to the hanging-wall contact.

9/ Thomas, Robert W., Mining Practices at Ray Mines, Nevada Consolidated Mining Co.: Bureau of Mines Inf. Circ. 6167, 1929, 27 pp.

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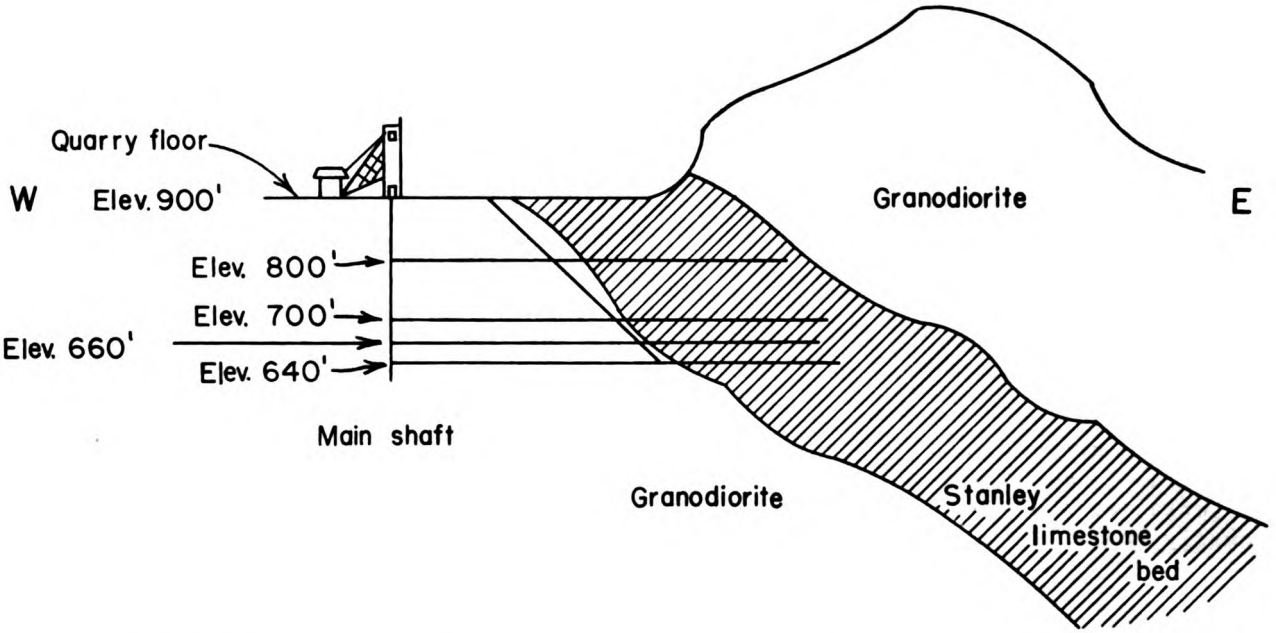
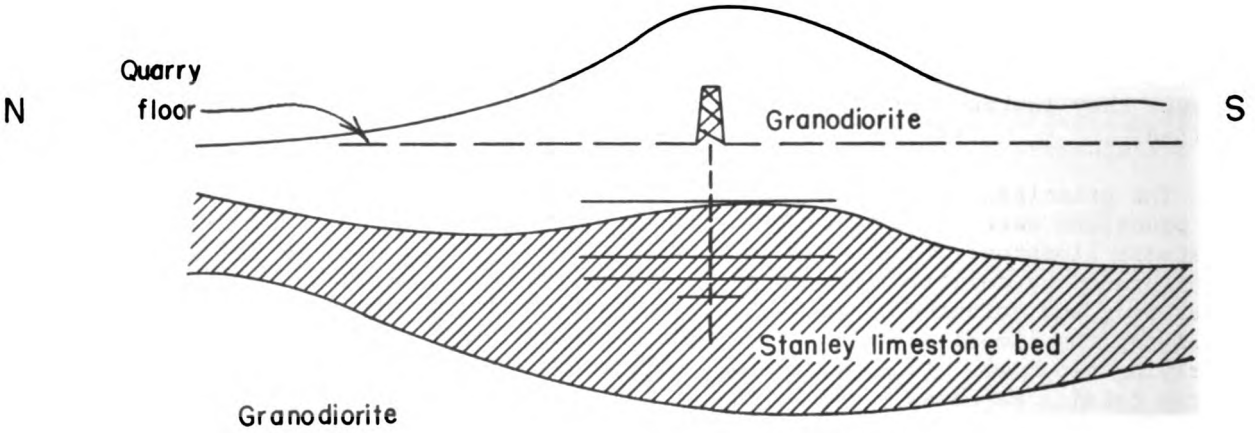


FIGURE 3. - East-West Section, Showing Placement of Principal Mine Workings in Relation to Lower Stanley Limestone Bed.



Elev. 800 = Manway and ventilation level
Elev. 700 = Grizzly and mining level
Elev. 660 = Haulage level
Elev. 640 = Sump and pumping level

FIGURE 4. - North-South Section, Showing Placement of Principal Mine Workings in Relation to Lower Stanley Limestone Bed.

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2. The area that undercut each block to start the caving was much greater than that undercut in normal block caving. At the Crestmore mine an undercut area about 80 by 200 feet (16,000 square feet) was generally required to initiate the cave, and as much as 35,000 square feet was undercut in one instance without producing a cave. In some porphyritic copper ores, caving can be induced by undercutting an area as small as 625 square feet.

3. To sustain caving, a radically different system of draw control was developed. In normal block caving the most widely accepted practice is to pull equal tonnage from each drawpoint according to a predetermined daily schedule. By following this procedure, funneling is minimized. At Crestmore no draw schedule was used, and pulls were confined to those drawpoints underlying the areas of the block highly enough stressed to fracture and break the rock. The area under high stress was usually indicated by the presence of a relatively copious amount of powdered rock in the drawpoint chutes. Pulls were made and continued only from drawpoints where the powdered rock was evident until a substantial void, possibly 75 feet in diameter, had been created. This arch over the void thus transmitted and concentrated the load to other localized areas in the block, and the pulling accordingly was shifted to the drawpoints, where the presence of powdered rock in the chutes indicated a high stress area. Once the draw was started from an area, it had to be continued at a rate sufficient to maintain a void. If the blocks were allowed to settle on the broken rock, it would reconsolidate, and a considerable effort would be required to reinitiate the cave. Sometimes reconsolidation was extensive enough so that the block had to be almost completely reundercut to restart the caving action.

The mining method developed at the Crestmore mine was characterized by a number of factors, some of which are seldom experienced or applied in mining porphyry-copper and other similar deposits by block-caving methods. Some of these factors that contributed to the success of the operation at Crestmore might be advantageously applied to other caving operations where the characteristics of the ore and country-rock materials are similar to those at Crestmore. These factors are:

1. The rock arched instead of funneling when a sustained pull was confined to a particular set of contiguous drawpoints.

2. Even though most of the daily production could usually be drawn from a single producing block, the amount pulled per day ranged from 700 to about 3,000 tons during the life of the mine. Under average caving conditions in other formations, as the porphyry coppers, this tonnage is a limited production. Factors limiting daily production in the Crestmore mine are:

(a) After an arched void develops, some time is required to allow the stresses in the block to readjust and for them to transfer and be concentrated to some area in the rock near the periphery of the void. The material in this peripheral area is overstressed and crushes, and some powdered rock is produced. This powdered rock filters downward through the broken zone toward the drawpoint chutes, where it accumulates. A considerable time may elapse before enough powdered rock accumulates in one or several drawpoints to become noticeable and hence serve as an indicator that the rock overlying these drawpoints is broken enough to be drawn. The readjustment period and the time spent in seeking the drawable chutes, as evidenced by the accumulated rock powder, are nonproductive.

(b) The draw must be confined to a restricted area of the block that can be pulled only from a few, usually contiguous, drawpoints.

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(c) The location of the drawable area shifts erratically. No set routing of mine cars to specific transfer raises is possible, and a continual, time-consuming rerouting of car trains is unavoidable.

3. About 30 percent of the limestone drawn was oversize. These oversize pieces were broken, either in the drawpoint chutes or on the grizzly level, by blockholing, mud capping, and bombs.

4. Difficulty was experienced in caving the last 25 percent of the block. This was especially true where the granodiorite cover was thin or nonexistent. Evidently the weight of the last 25 percent of the block and cover material was not great enough to cause good fragmentation.

5. The major portion of the granodiorite cover crushed to minus-1/4-inch material that dribbled down through the coarsely broken limestone blocks. As a result, the limestone was substantially diluted by the granodiorite. Dilution is a serious disadvantage in most mines, but at the Crestmore mine it was advantageous because 20 percent of a material containing alumina and silica was needed, plus the Stanley limestone, to make a product from which a portland cement could be produced. The granodiorite, which was drawn with the limestone as a diluent, served as the source of the needed alumina-silica material.

6. The strength of the limestone was such that little support was needed in either the haulage or grizzly levels. As the first blocks were developed, the grizzlies and drawpoint chutes were reinforced with concrete. After a period of operation this concreting was found to be unnecessary and was discontinued. The fact that little support and virtually no maintenance were required in the underground workings contributed substantially to the economic efficiency and safety of the Crestmore operation.

7. An advantage resulting from block caving in "hard" rock (limestone) was that all development work, the extraction of the pillars on the undercut level, could be performed well in advance of placing the blocks in production. Because developed blocks could be put into production on short notice by blasting out the undercut level pillars, the mine production could be matched to demand with virtually no delay.

8. A measure of mining efficiency is the production per man-shift. This could not be reconstructed over the entire life of the Crestmore mine, but from 1943 to 1951 a 30-man crew produced an average of 32.25 tons per man-shift. Daily production during this period averaged 967 tons. The mine crew consisted of 23 underground and 7 surface workers.

9. The safety record established at the Crestmore mine was exceptional. Between April 1941 and June 1953 the mine was operated without a single lost-time accident. Good management is generally the principal factor in achieving safety, but in underground work the stability of the ground is an element that cannot be ignored. As previously discussed under factor 6, the stability of the rock in the haulageways, cutoff stopes, undercut level, and particularly in the grizzly level contributed to the record of safety established at the Crestmore mine.

DETAILS OF BLOCK DEVELOPMENT AND CAVING

The sequence of block development and the date each block was put in reserve draw condition, withdrawn from active production, and finally closed down completely

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are shown in table 2. A minor amount of material was drawn from five of the blocks at infrequent intervals after they had been withdrawn from active production (see "Date of last draw," table 2).

TABLE 2. - Sequence in which blocks were developed and put in reserve draw condition

Block number	Date put on reserve draw	Date of last active production	Date of last draw
1A	May 1930	December 1941	December 1942.
2A	March 1931	do.	July 1944.
2B	December 1933	May 1948	-
1C	May 1935	December 1937	September 1939.
2C	July 1935	January 1938	-
1B	January 1936	December 1938	August 1941.
4A	May 1936	December 1941	September 1951.
3A	November 1937	January 1954	-
3B	do.	do.	-
4B	June 1938	August 1953	-
4C	June 1940	do.	-
5	January 1951	February 1954	-

A brief description of the development and the observed action of each block follows:

Block 1A

Block 1A was near the center and on the footwall side of the lower Stanley bed (fig. 5). The block was approximately 175 feet wide, 200 feet long, and 200 feet thick, as shown in plan and section in figures 6 and 7. Thickness of overburden (fig. 7) ranged from zero, where the limestone was exposed in the quarry floor, to 90 feet of granodiorite under the hanging wall.

Cutoff stopes on all 4 sides of the block (figs. 6, 7, and 8) were driven upward as shrink stopes in 2 lifts - from the manway level to the surface or contact with the hanging wall first and from the mining to the manway level last. The approximate dimensions of the cutoff shrinkage stopes are shown on the scaled sections (figs. 6, 7, and 8). At first the cutoff stopes were mined as narrow as possible, approximately 4 feet. This width was found to be too tight and impractical to maintain. By widening the stopes to about 8 feet, the overall powder consumption was increased only slightly, and mining efficiency was greatly improved.

Vertical pillars (X, Y, Z, and W, fig. 6) of unbroken limestone were left between the ends of the cutoff stopes at the four corners of the block. Fifty percent of the horizontal pillars (C and D, fig. 7) at the manway level were broken by blasting when the stopes driven upward from the mining level intersected the manway-level workings.

The cutoff-stope backs were reached through cribbed manways carried upward through the broken rock from the manway and mining levels. On completion the stopes were left full of broken rock. The stope fill material was eventually drawn out, along with the material pulled from the main body of the block after it was caved.

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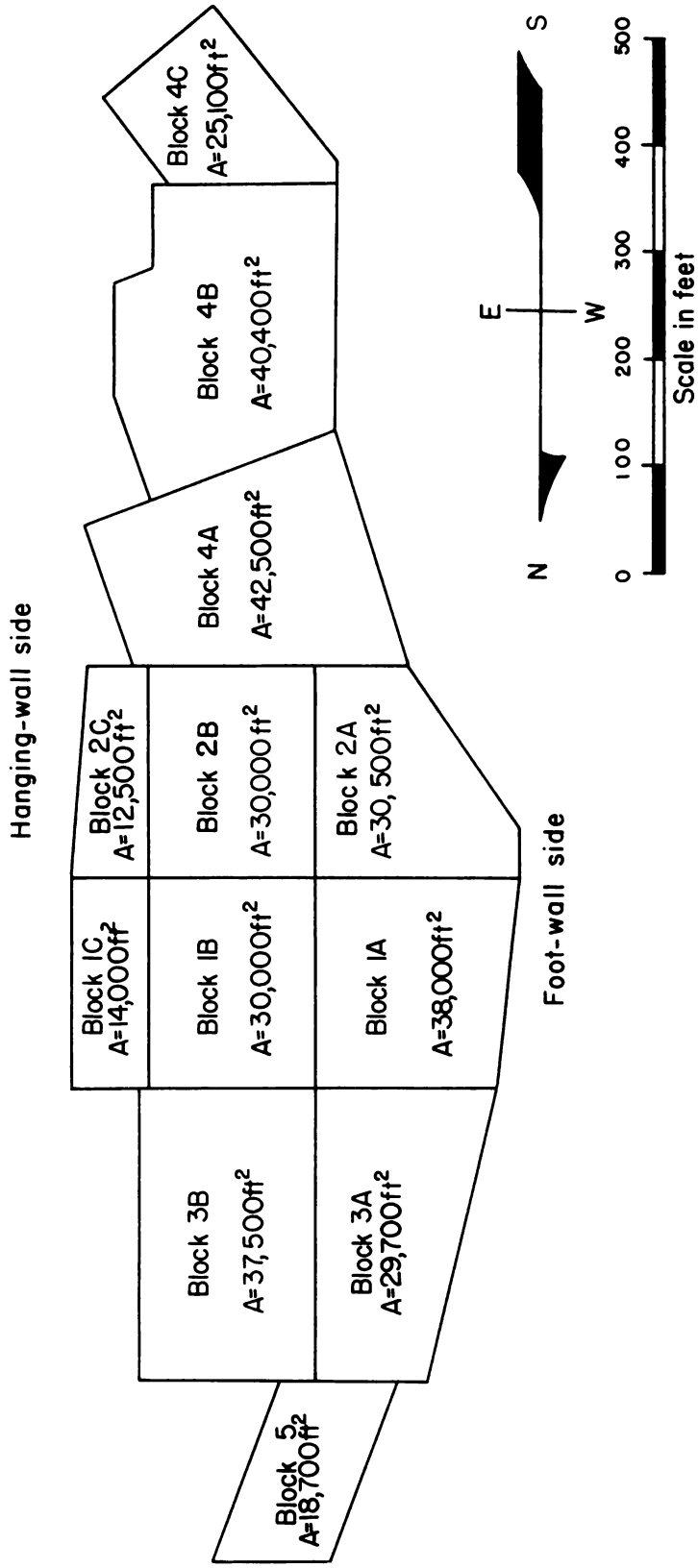

FIGURE 5. - Outline Plan of the Block Units Caved.

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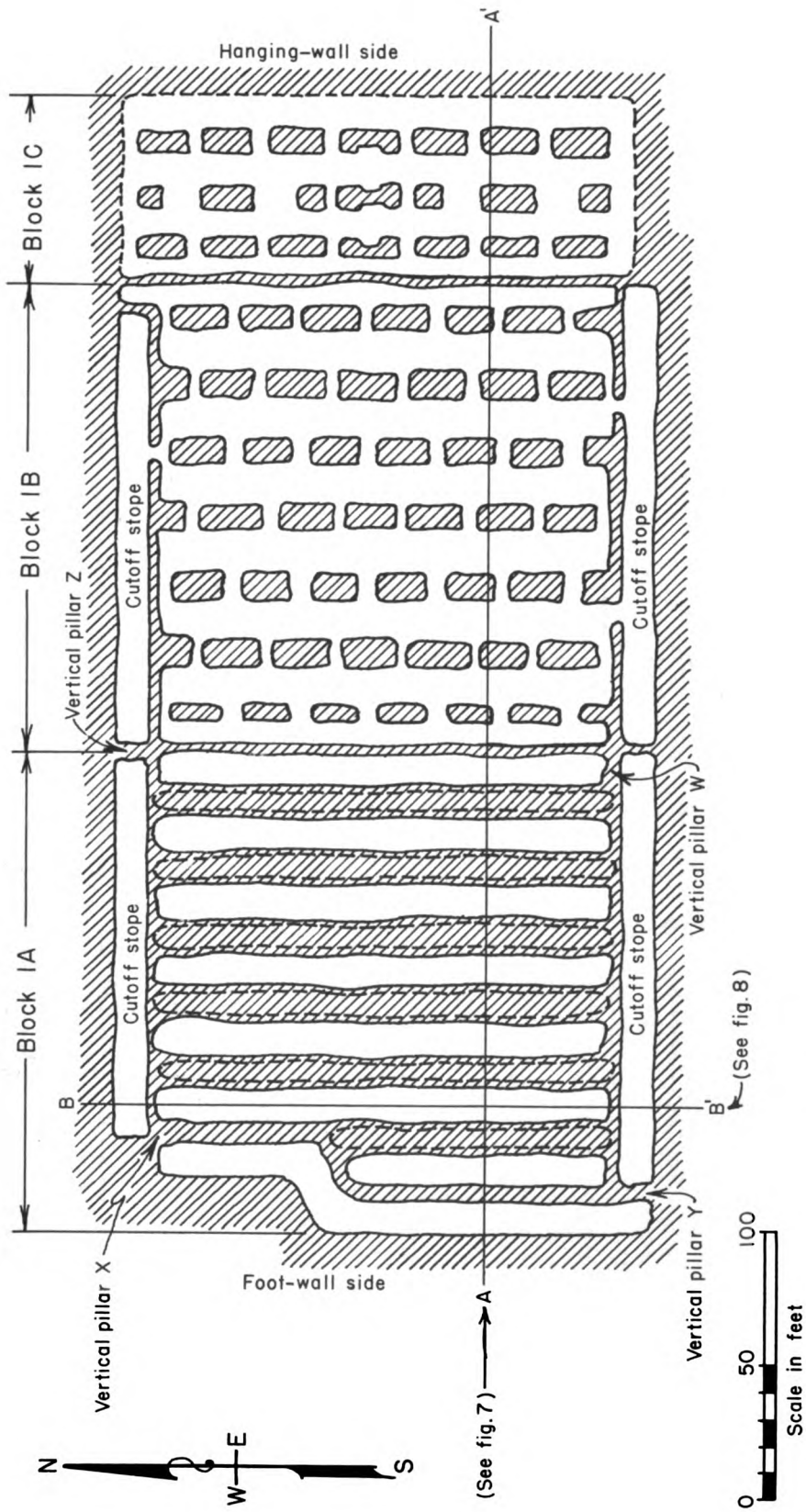

FIGURE 6. - Plan of Undercut Level of Blocks 1A, 1B, and 1C.

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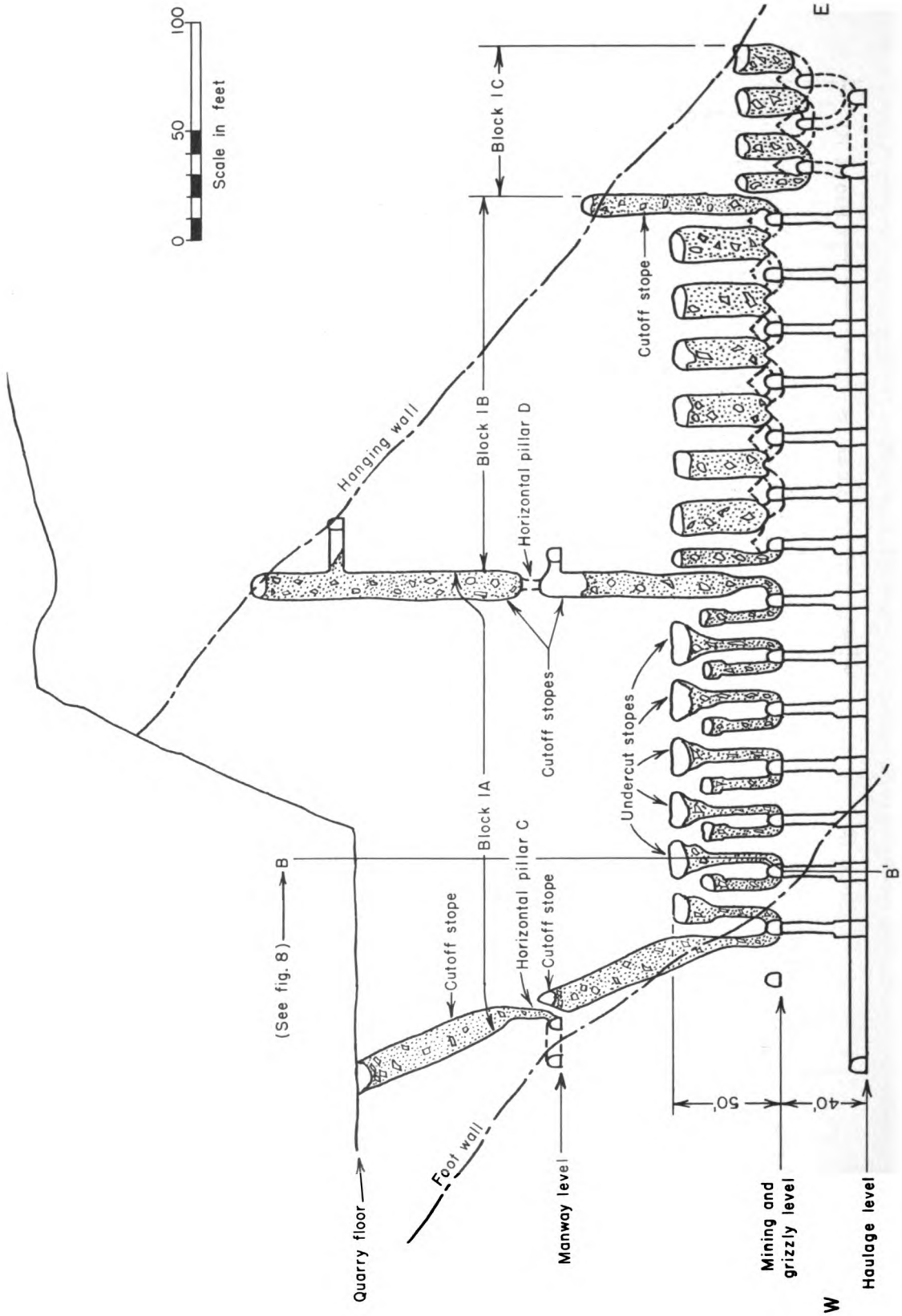

FIGURE 7. - Detailed East-West Section Through Blocks 1A, 1B, and 1C.

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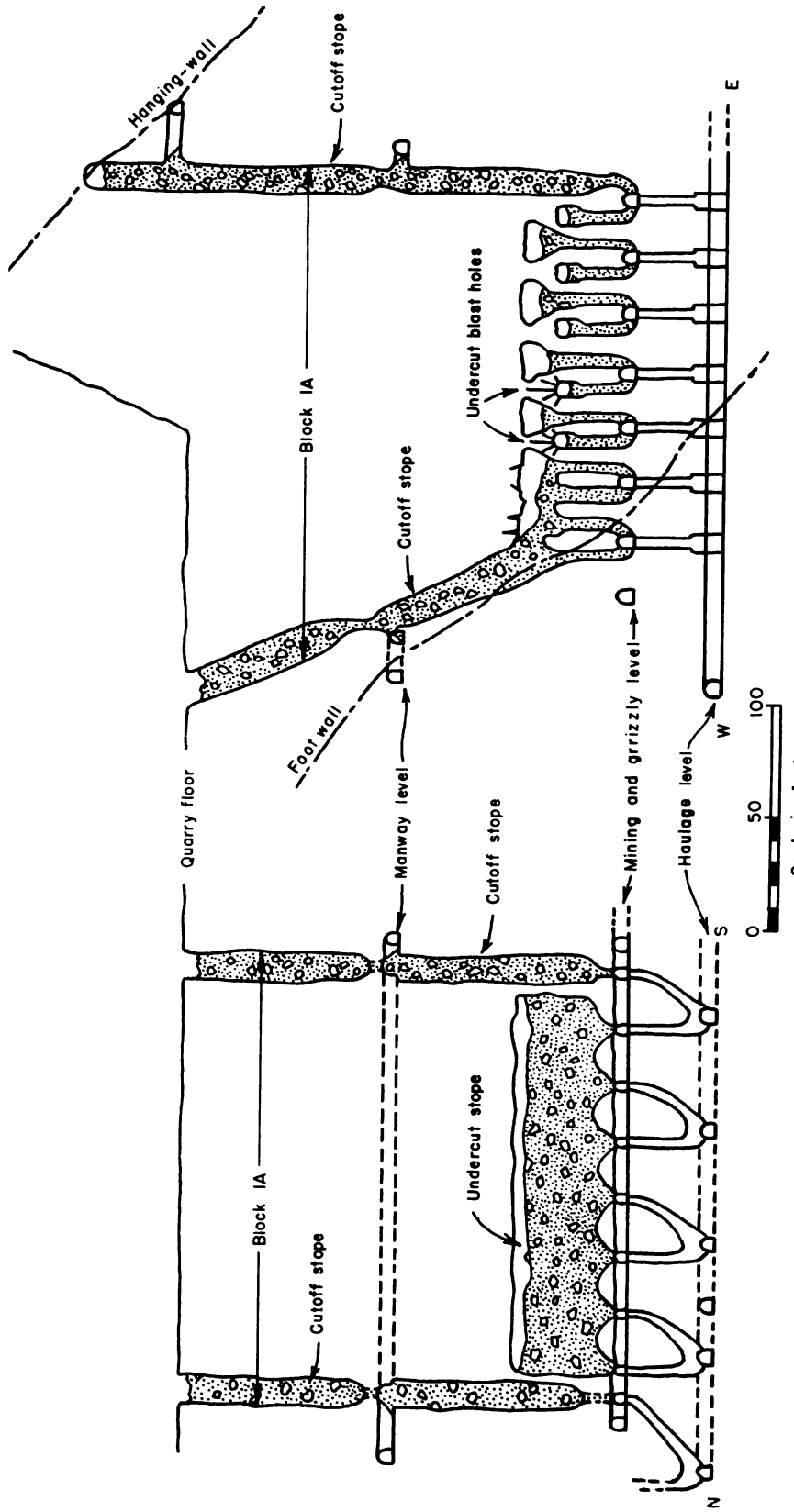

FIGURE 8. - Section B-B' Through Block 1A, Showing Cutoff and Undercut Stopes.

FIGURE 9. - Section A-A' Through Block 1A, Showing Initiation of Final Undercutting Operation.

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The fill left in the cutoff shrinkage stopes generally compacted enough so that it did not run excessively when the block was drawn. Conversely, fill compaction was not severe enough to hinder the draw from the main body of the blocks, once caving had been initiated. The fact that the fill would compact, especially when subjected to a localized transfer of stress at the periphery of a deliberately created, arched void, may have been a contributing factor in the tensional breakage of rock in the arches over the voided areas.

Block 1A was undercut by developing a series of parallel narrow stopes above the mining level, spaced about 25 feet apart. Figures 6, 7, 8, and 9 illustrate the undercut stope development in plan and sections. After all the undercut stopes under the block had been completed, the top of the pillar nearest the footwall was drilled, loaded with dynamite, and broken by blasting (fig. 9). This process was repeated on successive pillars from the foot wall toward the hanging-wall cutoff stope.

When block 1A had been completely undercut, it did not cave and remained in place until the two vertical corner pillars (Z and W, fig. 6) on the hanging-wall side were broken by blasting. The block then dropped "en masse" and was accompanied by an immediate breakthrough and subsidence of several feet at the surface. When the block dropped, there was no noticeable air blast in the underground workings; however, the desired caving fragmentation did not occur, and the block came to rest on top of the remnants of the undercut pillars, as shown in figure 10.

Some of the pillar remnants were cracked and somewhat shattered by the impact of the falling block. This pillar fracturing, combined with irregular spalling and rockfalls from the underside of the block, created a hazardous condition which persisted for some time. Two years elapsed before the pillar remnants could be removed safely or broken enough by blasting to induce a general caving action, as shown in figure 11.

Block 2A

Block 2A covered 30,500 square feet, and its height and overburden factors were similar to those of block 1A (see fig. 12).

Only 3 cutoff stopes were needed; 1 side was bounded by the caved area in block 1A. The vertical corner pillars between the ends of the cutoff stopes were reduced in size until they were virtually nonexistent in developing 2A and subsequent blocks.

Figure 13 shows, in plan, the size and distribution of the undercut pillars left under block 2A before caving was induced by completely breaking or removing the pillars by blasting. In this and subsequent undercutting operations the undercut pillars were loaded from top to bottom to insure that they would be completely broken and no projecting remnants would be left to interfere with subsequent caving and drawing procedures.

Starting from the footwall side, the shrinkage stope and successive rows of the undercut pillars were removed by blasting one row at a time. Three rows of pillars were removed before caving was induced. Hence, support under approximately 11,000 square feet of the block was removed before caving took place. Subsidence on surface did not occur immediately, as too much muck had been allowed to remain in the undercut area. When some of this muck was drawn, the cave worked itself to the surface in about a week.

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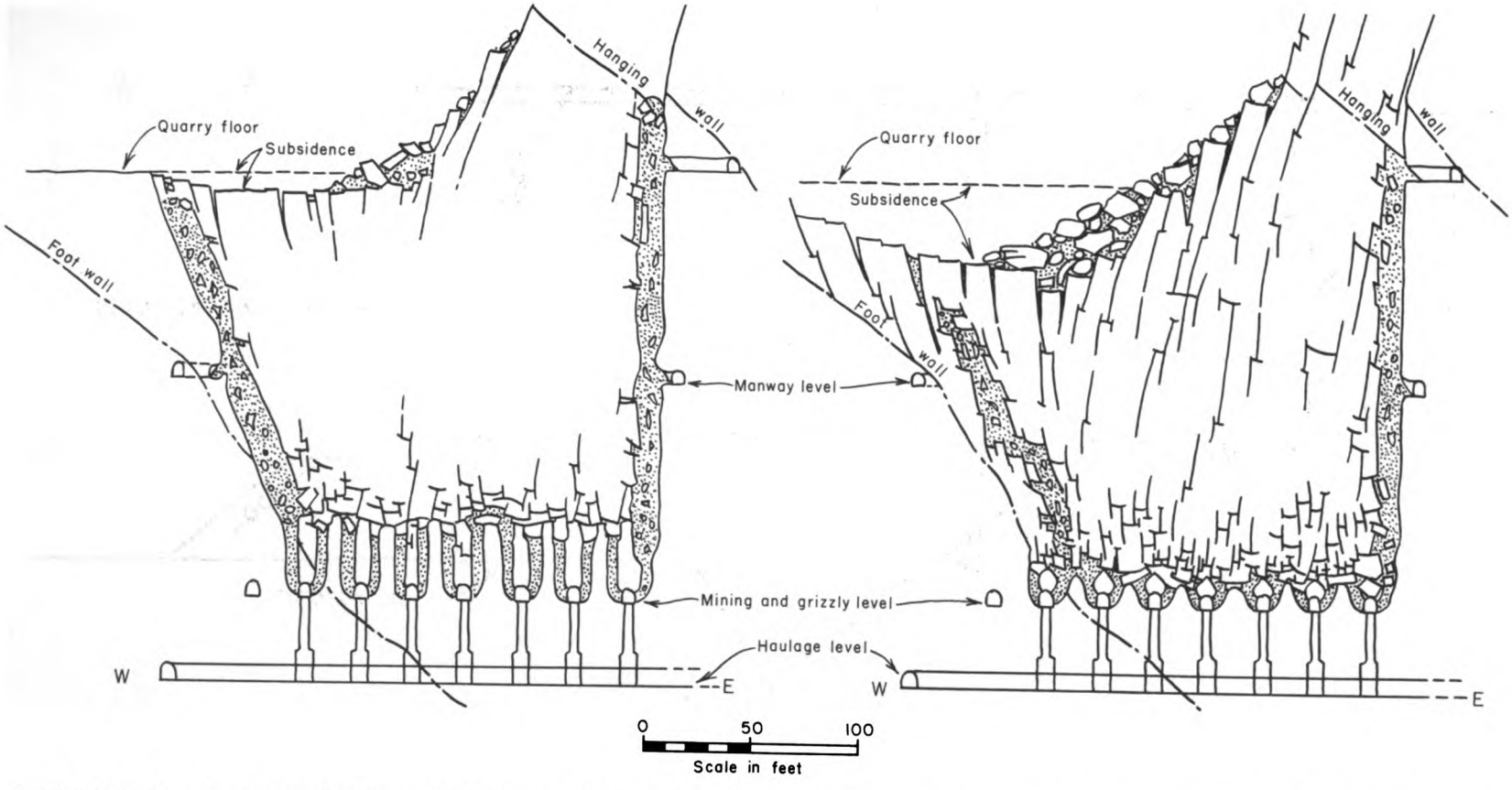

FIGURE 10. - Section A-A' Through Block 1A After Vertical Pillars Z and W Had Been Broken by Blasting.

FIGURE 11. - Section A-A' Through Block 1A, Showing the Initiation of Caving Action After Remnants of the Undercut Pillars Had Been Removed by Blasting.

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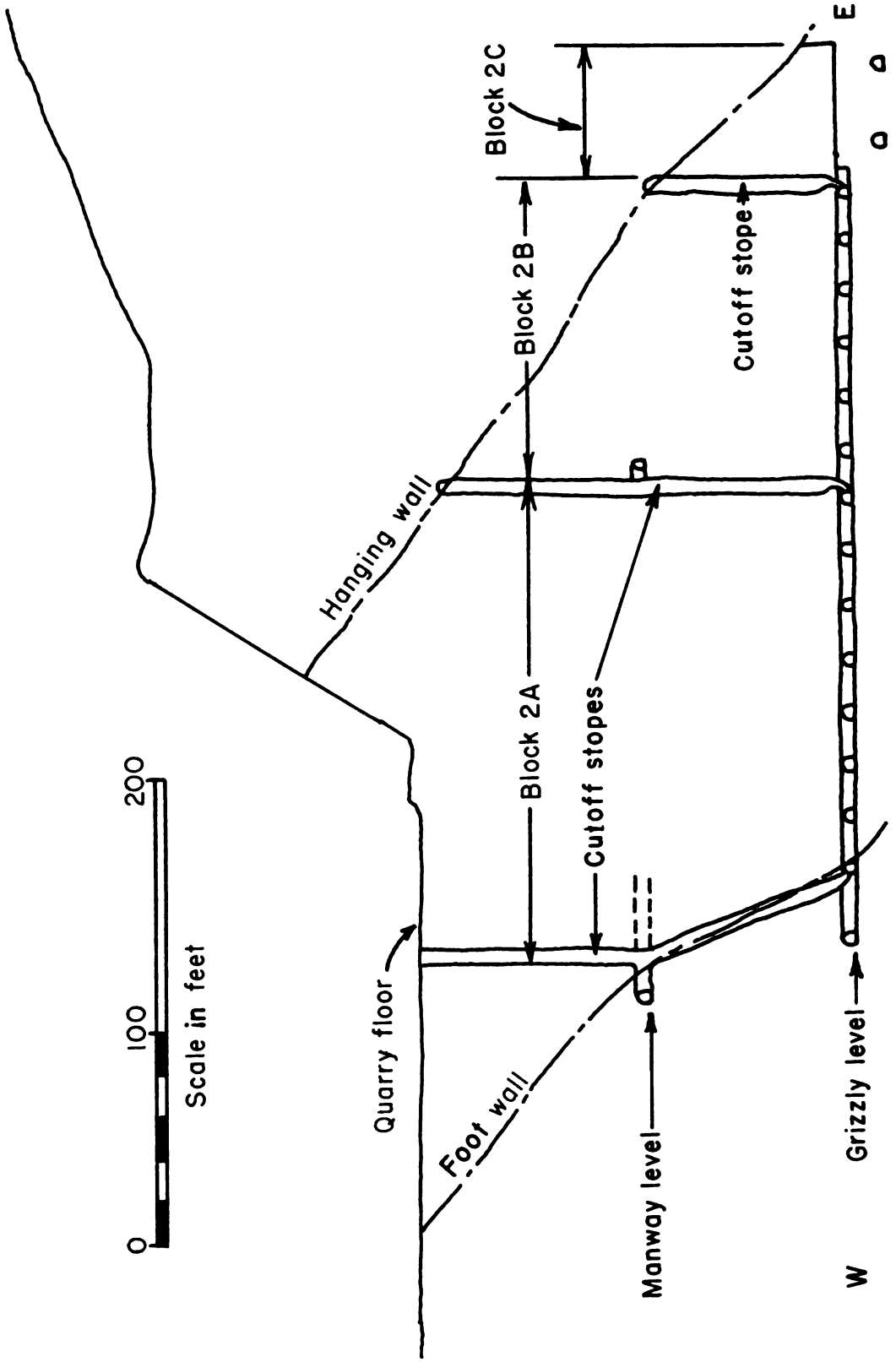

FIGURE 12. - Diagrammatic East-West Section Through Blocks 2A, 2B, and 2C.

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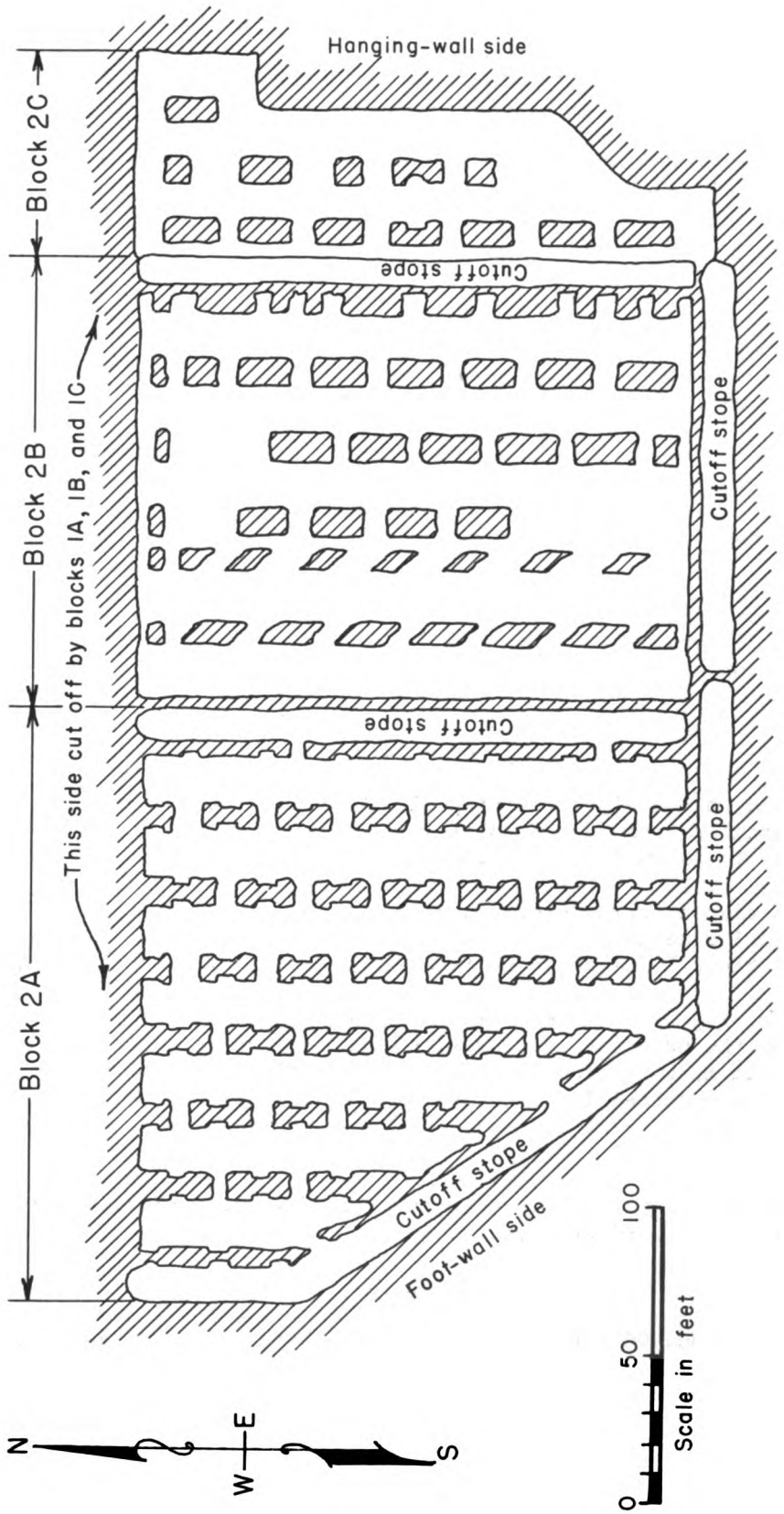

FIGURE 13. - Plan of the Undercut Level of Blocks 2A, 2B, and 2C.

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The remaining pillars were blasted, a row at a time, and caving action continued progressively. However, after the fifth row had been blasted out, too much muck was pulled, and excessive stress developed that entirely crushed the sixth row of pillars. The last row of undercut pillars and the stope pillars were removed in a single blast.

Surface subsidence was progressive, and no piston effect was noted because the block never dropped "en masse."

Block 2B

Block 2B covered 30,000 square feet. Overburden consisted of granodiorite ranging from 130 to 270 feet in thickness (see fig. 12). Hence, none of the cutoff stopes intersected the quarry floor. The pattern and distribution of the undercut pillars differed somewhat from those under block 2A, as shown in figure 13; but the mining scheme was not changed materially.

Again removal of the undercut pillars proceeded from the foot toward the hanging wall, and 3 rows, or about 16,000 square feet, was undermined before caving began. These first 3 rows were removed within 3 weeks. During the next few weeks the remaining undercut pillars were removed a row at a time.

When the undercut pillars were being removed, draw was carefully controlled to keep a mat of broken material in contact with the subsiding block, and none of the remaining rows of pillars were crushed by a buildup of excessive stress. Hence, surface subsidence progressed at an even rate from the area previously caved by block 2A.

Blocks 2C and 1C

Block 2C was a narrow block covering 12,500 square feet. Overburden consisted of granodiorite ranging from 290 to 340 feet in thickness (see fig. 12). A plan of the undercut level is shown in figure 13.

Block 1C was also a narrow block covering 14,000 square feet. Overburden consisted of granodiorite ranging from 290 to 380 feet in thickness. A plan of the undercut level of block 1C is shown in figure 6 and a vertical section in figure 7.

No cutoff shrinkage stopes were mined on the end boundaries of blocks 1C and 2C, but stoping on the undercut level was extended upward to intercept the granodiorite hanging wall in both blocks. After undercutting under both blocks was completed, no caving occurred in either block for approximately 6 months; then both blocks dropped "en masse", with an immediate breakthrough to and subsidence on the surface.

Much trouble was encountered in attempting to draw from these blocks, and all the limestone was never completely recovered.

Block 1B

Block 1B covered 30,000 square feet, with overburden consisting of granodiorite ranging from 100 to 280 feet in thickness. A plan of the undercut level is shown in figure 6 and a vertical section in figure 7.

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The development scheme followed the pattern established in blocks 2A and 2B, except that no horizontal pillars were left in the shrinkage cutoff stopes at the manway level. However, in undercutting block 1B removal of the rows of undercut pillars proceeded from the hanging toward the foot wall sides of the block. Five rows of pillars were removed, leaving 26,000 square feet of the block unsupported, but no caving occurred. The remaining two rows of pillars were blasted simultaneously, and the entire block dropped as a unit. Breakthrough to surface, accompanied by several feet of subsidence, was concurrent with the blast that shattered the last two rows of undercut pillars. The impact of the dropping block was strong enough to crush the roof pillar over one of the grizzly level drifts, and the drift was lost.

Considerable difficulty was experienced in starting and maintaining draw from block 1B.

Blocks 3A, 3B, and 4A

Blocks 3A, 3B, and 4A were mined in the same way as block 2B, and in each instance about the same amount of undercutting was required before caving was begun. In all three blocks caving and subsidence were progressive, and no particular difficulty was experienced in drawing.

Blocks 4B and 4C

As shown in figure 14, the undercut pillars were changed to a rib instead of a grid of rectangular pillars. This system of undercutting was devised to overcome the company's inability to obtain experienced miners during the war years. However, it proved to be a safer and more easily controlled system regardless of whether or not the miners were experienced.

In both blocks the footwall cutoff stopes were emptied, and part of the rock on the quarry floor was drawn through the empty shrinkage stope after having been broken by underhand benching methods. Removal of the rib pillars was started at the foot wall with retreat toward the hanging wall side of the block. From 12,000 to 16,000 square feet of the blocks was undermined before caving was induced. Subsidence was progressive, and no difficulty in drawing was experienced except when insufficient labor curtailed the draw for too long a time.

Block 5

Block 5 was in an area in which numerous large, irregular solution cavities and crevices were encountered. Hence, the cutoff stopes on the footwall and north ends of the block were not completed, and placement of the undercut pillars followed no set pattern. Undercutting with a satisfactory cave action was accomplished without difficulty, though the block was never completely drawn, as most of the material needed by the cement mill was obtained from development of the room-and-pillar mine.

The "last chapter" to this report remains to be written. This chapter will compare the relative efficiency or cost of block caving with open stoping at Crestmore. Preliminary figures already indicate that open stoping will show a substantial advantage over block caving.

In retrospect, it might be asked why block caving was originally chosen over open stoping. Even though the upper 200 feet of the limestone was weakened by solution channels, its strength should not have excluded mining by open stoping.

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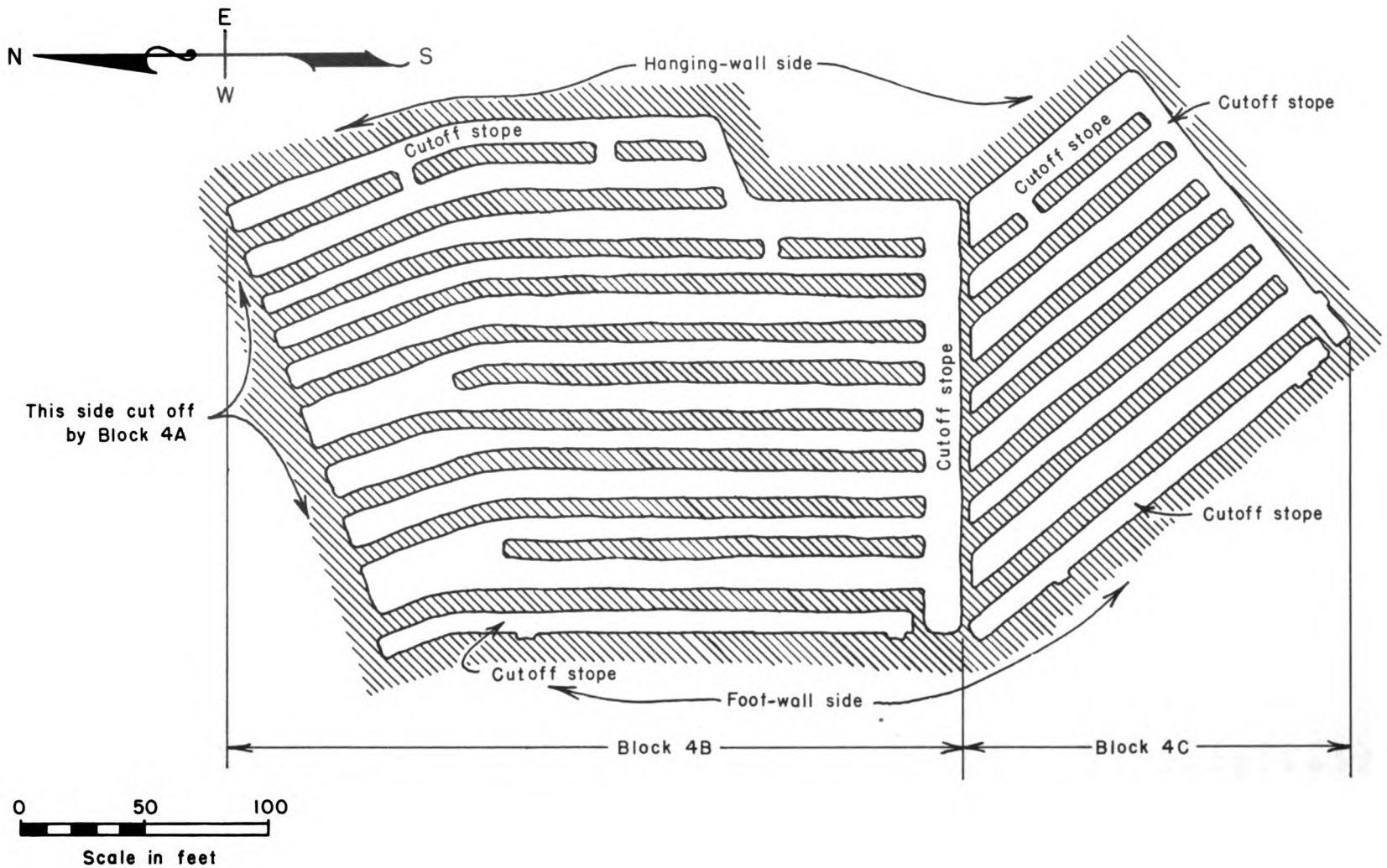

FIGURE 14. - Plan of Undercut Level of Blocks 4B and 4C.

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The factor that now favors open-stoping is the improved, large-scale, efficient, underground mining equipment that was not available 25 years ago.

CONCLUSIONS

Block caving in "hard" rock as practiced at the Crestmore mine has several advantages over conventional caving in the porphyry-copper deposits.

1. The block development could be made well in advance of putting the block into production, thereby allowing flexible mine production.

2. The method was satisfactory for a relatively low daily production. The draw from 1 block could be maintained at a rate as low as 700 tons per day.

3. Timber and other support in haulage, grizzly, and undercut levels and in shrinkage stopes were generally unnecessary.

4. Good efficiency was achieved, as indicated by an 8-year average production of 32.25 tons per man-shift.

5. Stable ground conditions contributed to achievement of an exceptional safety record.

The method has the following advantages:

1. A large proportion of the draw was oversize. About 30 percent of the total draw had to be broken in the chutes or on the grizzlies by secondary blasting.

2. Considerable difficulty was experienced in recovering the final 25 percent of each block because the internal stress in the remnant block resulting from the overlying rock and cover was too low to cause the block to fragment.

3. Considerable dilution from the cover must be expected, especially during the final phase of the draw from the block.

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